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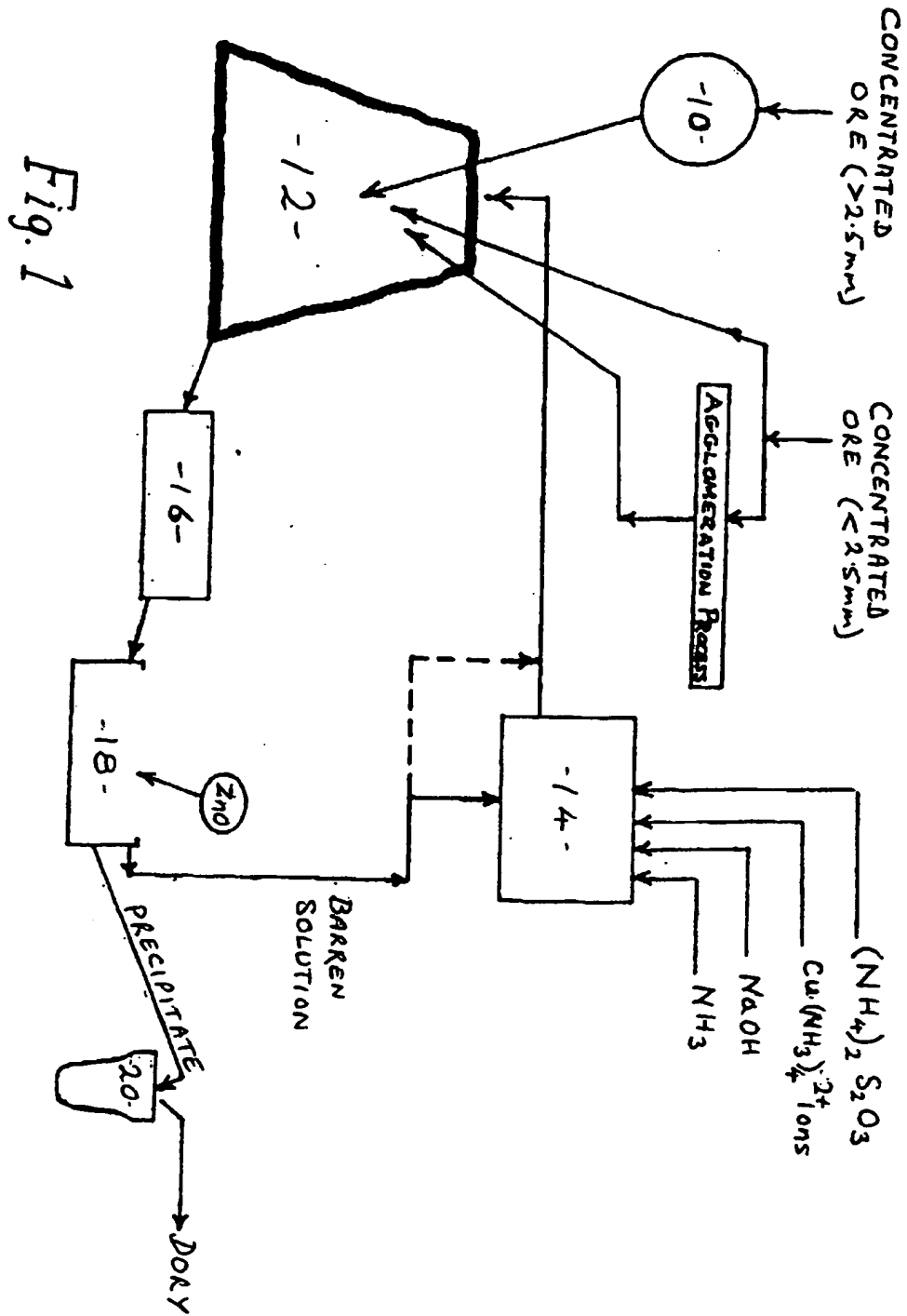
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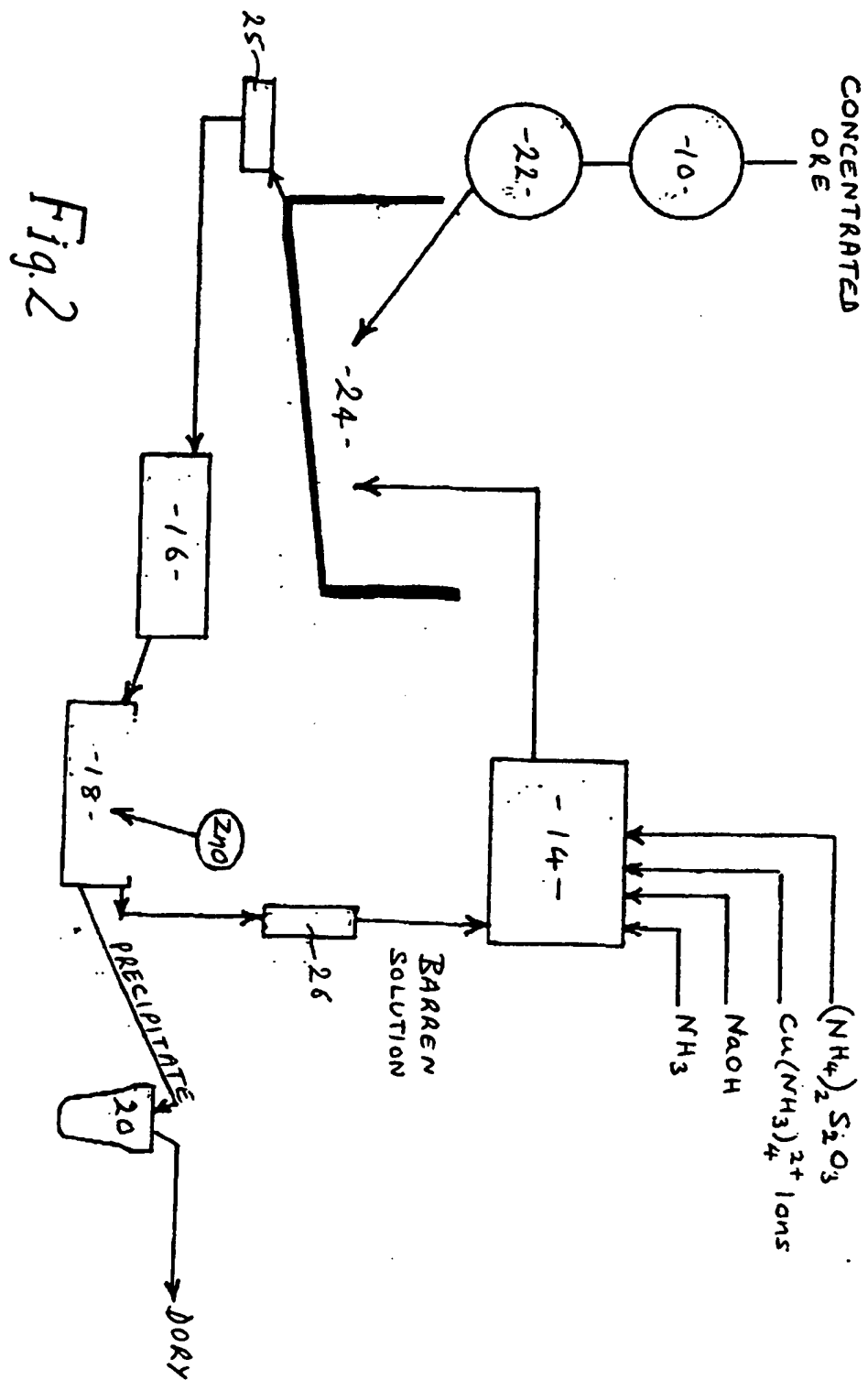
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(54) Recovering gold from oxide ores

(57) A method for recovering gold from oxide ores containing 0.5 to 7.5 weight % of carbonaceous materials comprises leaching the concentrated ore with an alkaline thiosulfate (eg ammonium thiosulfate) solution to which an oxidizing agent and a stabiliser are added, gold being separated from the resulting saturated slurry by a zinc cementation precipitation method. The oxidizing agent may be H₂O₂, FeCl₃ or, preferably, cupric tetrammine ions. The stabiliser may be bisulfite or ammonia.

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PROCESS FOR RECOVERING GOLD FROM OXIDE-BASED
REFRACTORY ORES

This invention relates to a metallurgical process for the recovery of gold from concentrated oxide refractory ores containing gold and carbonaceous materials by leaching with a thiosulfate solution and precipitating gold from the pregnant leaching slurry using a zinc cementation method.

The leaching portion of the process is designed to be effective for both low grade ores using a heap method of percolation, and for high grade ores by agitating the pulp in a vat or tank.

Typically, gold is extracted from its ore by leaching with cyanide solution. However, refractory ores, those containing organic carbonaceous matter impurities, cannot be effectively processed using the customary cyanide method because the carbonaceous matter adsorbs gold dissolved in the leaching solution. This carbon-adsorbed gold is then lost in the waste stream, thus leading to poor or even zero gold recovery from the entire leaching operation.

Although the carbonaceous material can be pretreated so as to reduce its capacity to adsorb gold, such treatments, be they flotation, chlorination, roasting, chemical or biological, have not been entirely effective. Further, they require additional operations and add extra costs and complexities to the extraction process. (See US-A-5,127,942 which describes a microbial process to

deactivate the gold adsorbing properties of carbonaceous materials; and US-A-4,801,329 which uses a chemical oxidation pretreatment for the same purpose).

Where precious metal bearing ores have been unresponsive or difficult to treat with cyanide, the use of a thiosulfate leach, in lieu of cyanide, has been effective for non-carbonaceous ores. (See US-A-4,654,078; 4,369,061; 4,269,622 and 4,070,182).

A thiosulfate lixiviant has also been demonstrated to be successful on low grade refractory sulfide gold ores without the need to pretreat the organic carbonaceous matter they contain. (See US-A-5,354,359).

However, prior art does not establish that high recovery yields of gold can be obtained from oxidised refractory ores, both high and low grade, by using a thiosulfate leaching solution and without the need for pretreatment to remove or deactivate gold adsorbing carbons.

An object of this invention therefore is to provide a process for recovering gold from carbonaceous oxide ore, specifically those oxide ores containing between 0.5% and 7.5% of organic carbon, without the need of pretreating the ore to eliminate or deactivate the gold adsorbing carbon component of the ore.

A further object of this invention is to apply the process to both high and low grade oxidised refractory ores including ores that previously were considered waste

because their gold content was too low to permit economically feasible recovery.

The present invention provides a hydrometallurgical process for the recovery of gold from oxide ores containing precious metals and organic carbonaceous materials. The process consists of:

1. Proper preparation of the pre-concentrated ore.
2. Drenching the prepared ore with, or in, a certain thiosulfate solution under conditions that will rapidly saturate the solution with gold extracted from the ore.
3. Recovering the gold from the solution in a manner that results in extremely high yield in a short period of time.

In preparation for the first step the mined ore must be concentrated. The degree of concentration will depend largely upon the gold content of the ore, and the nature of the deposit. Generally the concentration will be done by conventional means - gravitational methods, flotation, roasting or some combination of these methods.

This invention comprises a process in which several steps may vary depending upon whether the ore being processed contains a high or a low concentration of gold. For the purposes of this invention we are defining low grade ore as that oxidised ore which contains 0.5% or more of organic carbonaceous material by weight, and less than four grams per metric tonne of gold. For such ores the

process followed is that described below for low grade ores.

Similarly, for the purposes of this invention we define high grade ore as that oxidised ore which contains 0.5% or more of organic carbonaceous material by weight, and six grams or more of gold per metric tonne. For such ores the process followed is that described below for high grade ores.

For those oxidised carbonaceous ores containing between four and six grams of gold per metric tonne either process may be followed. For such ores the process selected will be determined more by economic considerations such as the characteristics and peculiarities of the ore body, and its location, than by the gold content of the ore *per se*.

Preferred embodiments of the invention will now be described with reference to the accompanying drawings wherein:

FIG. 1 is a schematic representation of a process in accordance with a first embodiment of the invention for recovering gold from low grade ores; and

FIG. 2 is a schematic representation of a process in accordance with a second embodiment of the invention for recovering gold from high grade ores.

Referring to Fig. 1, a process for low grade ores comprises the following steps:

Step 1: The first step of the process is simply to crush

or grind the dry concentrated ore in a crusher 10. The particle size desired will depend upon the leaching method selected to be used in step number 2. Generally, when dealing with low grade ore the leaching process will be done in a static heap 12. In such an event, the ore will be crushed to particles of approximately 20-25 mm in size, usually in a cone crusher.

It should be noted that some ores, especially weathered crusts or sand and gravel placers, may already be composed of particles of an appropriate or smaller size, and thus not require crushing. For such ores step number 1 may be omitted.

It should also be noted that some ore deposits consist of particle sizes so small that they will tend to compact, thereby reducing the porosity of the static heap. Such a condition results in inadequate surface area of the ore being exposed to the leaching solution. This prolongs the leaching operation, increases costs, and significantly reduces gold yield.

To overcome such a condition such ores require agglomeration prior to the leaching process. The procedure for doing this is:

- a) Add portland cement to the ore, approximately 2.5 kg per metric tonne of ore. Mix with water in sufficient quantities to form a stiff, barely viscous, moist mass.
- b) Deposit this material on an upwards running

conveyor belt. It will form into small balls usually about 5 cm or less in diameter.

c) Let the balls flow off the conveyor to form the heap 12.

d) Let it cure for two to three days before commencing the leaching operation..

Step 2: In a mixing tank 14, prepare a water-based leaching solution, preferably of ammonium thiosulfate to a concentration of approximately 20-25 grams of thiosulfate per litre of solution. Use sodium hydroxide to maintain a pH in the range of approximately 9.5-10.5, preferably 10.0-10.3

Step 3: Add an oxidizing agent such as hydrogen peroxide, ferric chloride or preferably cupric tetrammine ions, $\text{Cu}(\text{NH}_3)_4^{2+}$, in a concentration of 20-30 ppm.

Step 4: Add a stabiliser to the leaching solution in the form of bisulfite (HSO_3^-) or preferably ammonia, generally in a concentration of at least 1.0 ppm and preferably 2.0 ppm.

Step 5: Pass the thoroughly mixed leaching solution prepared in Steps 2 to 4 above through the heap 12 of crushed concentrated ore. Use the conventional drip method: PVC or high density polyethylene piping with a line pressure of 10-20 psi (i.e. 0.7-1.4 kg/cm²) to pressure drip emitters with not less than one emitter per square meter of top surface area of the heap.

Apply the leaching solution at a rate of 12 to 20

litres per minute per 100 sq. meters of heap top surface area (i.e. not less than 12 and preferably 20 litres per 100 sq. meters). Note: After the heap has become completely wetted, the delivery rate of thiosulfate solution can be cut to about 10-12 ℓ/min per 100 sq. meters if desired.

Step 6: Continue the flow of leaching solution through the heap for 4 to 20 hours. Typically the time required to maximise the gold ions entering solution will be 4 to 8 hours depending upon such factors as the size (volume) and porosity of the heap, the flow rate of the thiosulfate solution, how finely the ore has been ground, and the slurry temperature (which should be maintained at 30-40°C for optimum results).

Step 7: Draw off the saturated slurry from the bottom of the heap, either continuously or intermittently, into a "pregnant" solution storage tank 16.

Step 8: Separate the gold from the saturated slurry using the zinc cementation precipitation method in a tank 18.

Note: In the Merrill-Crow zinc cementation process it is necessary to de-aerate the saturated cyanide solution prior to cementation to avoid excessive zinc consumption and incomplete recovery of gold. But by using a thiosulfate solution with oxidizer and stabiliser as described above, instead of a cyanide leaching solution as is done with Merrill-Crowe, the need for de-aeration is avoided.

The amount of zinc powder required for maximum

precipitation will vary with the gold content of the solution and its temperature. Generally, 0.25-0.30 grams of zinc dust per litre of thiosulfate solution containing under 2.0 ppm of gold, will yield 100% recovery in about ten minutes at 25°C.

Step 9: Smelt the precipitate in a smelter 20 to remove the zinc. The residue, upon cooling, will be a "dory", typically an alloy of gold and silver. The gold content of the dory may be separated using any of the customary methods:

- a. Treating with nitric acid or hot sulfuric acid to dissolve the silver.
- b. Chlorination of the molten metal, and removing the resulting silver chloride either as fumes or by skimming.
- c. Electrolysis with steel wool as the cathodes.

Step 10: Recirculate the now "barren" leaching solution to the top of the heap 12 for reuse (unless testing indicates it is out of specification in which case return it to the mixing tank 14 for rebalancing).

A process for high grade ores will now be described with reference to Fig. 2.

Instead of being heap-processed, high grade gold ores will generally be treated by mixing and stirring the thiosulfate solution with the ore in a tank or vat 24. Using this technique, porosity of the ore is no longer a consideration, and the particle size can be much smaller to

facilitate more complete and more rapid interaction with the leaching solution.

The steps for processing high grade ores as defined in this invention are:

Step 1: Grind or crush the dry concentrated ore to a 74 μm (200 mesh) size, for example using a crusher 10 and/or a grinder 22.

Step 2: Prepare the thiosulfate leaching solution and mix in a tank 14 with the oxidizing agent and stabiliser as described above in steps number 2, 3 and 4 for low grade ores.

Step 3: Place the finely ground ore in a metal or concrete tank or vat of appropriate size. Preferably this will be an open top reinforced concrete tank 24 with sloping floor and very smooth, or fibreglass-covered sidewalls and floor.

Step 4: Force under pressure the leaching solution into the tank 24 containing the ore in such a manner as to set up counter currents within the tank. Maintain the solution temperature at 30-40°C. Use power driven paddles extending into the slurry to constantly stir and mix the slurry. Add extra thiosulfate solution as needed to maintain proper consistency. Typically, the mix will require 150-400 litres (usually about 300 litres) of solution per metric tonne of ore in the leaching tank.

Step 5: After 2 to 8 hours, but typically 4 hours, draw off the saturated slurry through filters 25 at the bottom

of the leaching tank. The saturated slurry may be pumped to a "pregnant" solution tank 16 for temporary storage, or directly to the gold separation tank 18 in which a zinc cementation method is used to precipitate the gold contained in the saturated solution. This is done in the same manner as described above in step number 8 of the process for low grade ores.

Step 6: Smelt the precipitate in a smelter 20 to remove the zinc. (Sell the resulting alloy in dory form to a refiner, or continue with one or more gold separation methods as described in step number 9 of the process for low grade ores).

Step 7: Filter again and recirculate the now barren thiosulfate leaching solution to the solution mixing tank. Add replacement quantities of ammonium thiosulfate, oxidizer, stabiliser, water, and ammonium hydroxide as needed to balance the solution to prescribed content and pH. The now reconstituted leaching solution can be reused for repeat applications of step 4 above.

Step 8: Remove and dispose of the waste material at the bottom of the leaching tank either by scraping, lifting out with a front-loader, or preferably by flushing down the waste drain with a high volume of water into a holding pond.

CLAIMS:

1. A method of recovering gold from an oxide ore containing gold and 0.5 to 7.5% by weight of organic carbonaceous materials, comprising the steps of leaching the concentrated ore with a thiosulfate leaching solution to form a saturated slurry and separating the gold from the slurry by a zinc cementation precipitation method, characterised in that the leaching solution is maintained at alkaline pH and an oxidising agent and stabiliser are added thereto.
2. A method according to claim 1 wherein the leaching solution contains ammonium thiosulfate as a lixiviant.
3. A method according to claim 1 or claim 2 wherein the thiosulfate concentration in the leaching solution is 15 to 30 g/l.
4. A method according to claim 3 wherein the thiosulfate concentration in the leaching solution is 20 to 25 g/l.
5. A method according to any preceding claim wherein the pH of the leaching solution is from 9 to 11.
6. A method according to claim 5 wherein the pH of the leaching solution is 9.5 to 10.5.
7. A method according to claim 6 wherein the pH of the leaching solution is 10 to 10.3.
8. A method according to any preceding claim wherein the oxidizing agent comprises hydrogen peroxide, ferric

chloride or cupric tetrammine ions.

9. A method according to claim 8 wherein the leaching solution comprises 20 to 30 ppm of cupric tetrammine ions.

10. A method according to any preceding claim wherein a bisulfite or ammonia is added to the leaching solution as a stabiliser.

11. A method according to claim 10 wherein the stabiliser comprises at least 1 ppm of ammonia, preferably at least 2 ppm.

12. A method according to any preceding claim wherein the oxide ore comprises 6 g/tonne or less of gold, and wherein the leaching is carried out by passing the leaching solution through a heap of the concentrated ore.

13. A method according to claim 12 wherein during the leaching step the ore has a particle size of 10 to 40 mm.

14. A method according to claim 13 wherein the average particle size of the ore during leaching is 20 to 25 mm.

15. A method according to any one of claims 12 to 14 wherein during the leaching step the leaching solution is dripped through the heap of ore at a rate of 12 to 20 litres per minute per 100 square metres of top surface area of the heap.

16. A method according to claim 15 wherein the leaching step is carried out for a period of 4 to 20 hours.

17. A method according to any one of claims 12 to 16 wherein the leaching step is carried out at a slurry temperature of 30 to 40°C.

18. A method according to any one of claims 1 to 11 wherein the ore comprises 4 g/tonne or more of gold and the leaching step is carried out by mixing and stirring the leaching solution with the ore in a tank.

19 A method according to claim 18 wherein for the purposes of the leaching step the dry concentrated ore is ground or crushed to a particle size of 50 to 100 μm .

20. A process according to claim 18 or claim 19 wherein the leaching solution is forced under pressure into the tank containing the ore, in such a manner as to set up counter currents within the tank.

21. A method according to any one of claims 18 to 20 wherein 150 to 400 litres of leaching solution per tonne of ore are added to the leaching tank.

22. A method according to any one of claims 18 to 21 wherein the slurry is mixed in the tank for a period of 2 to 8 hours.

23. A method according to any one of claims 18 to 22 wherein the leaching slurry is maintained at a temperature of 30 to 40°C.

24. A method of recovering gold from a low grade oxide ore, substantially as herein described and illustrated with reference to Figure 1 of the accompanying drawings.

25. A method of recovering gold from a high grade oxide ore substantially as herein described and illustrated with reference to Figure 2 of the accompanying drawings.



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Claims searched: 1-25

Examiner: C.A. Clarke
Date of search: 14 May 1997

Patents Act 1977
Search Report under Section 17

Databases searched:

UK Patent Office collections, including GB, EP, WO & US patent specifications, in:

UK CI (Ed.O): C1A (AP31,AP32,AP33,AP34,AP37,APDT)

Int CI (Ed.6): C22B 3/14,11/00

Other: ONLINE: WPI

Documents considered to be relevant:

Category	Identity of document and relevant passage	Relevant to claims
A	US 5536297 MARCHBANK	
A	US 4269622 KERLEY	

X Document indicating lack of novelty or inventive step
Y Document indicating lack of inventive step if combined with one or more other documents of same category.

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A Document indicating technological background and/or state of the art.
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